A process is disclosed for the suspension or flash smelting of finely-divided oxide and sulfide ores and concentrates, a process in which a suspension of a finely-divided raw material in pre-heated air or oxygen is fed downwards at the reaction temperature or a higher temperature in a reaction zone formed together by the suspension and the smelt underneath in order to oxidize and partially smelt the suspension raw material, whereafter the oxidized raw material is partially sulfidized and the suspension flow is made to change its direction perpendicularly to the side so that the bulk of raw material in the suspension flow impinges towards the surface of the accumulated smelt in the main smelt reaction zone under the suspension reaction zone, and the remaining suspension flow is fed to a zone which has a rising flow, where the suspension is possible after-sulfidized and cooled and the solids to be returned to the suspension reaction zone are separated from the residual suspension flow.

The suspension flow portion not impinging against the smelt surface in the main smelt reaction zone is fed into a secondary reaction zone wherein it is allowed to, at least, partially dissolve in the smelt before the undissolved residual suspension is fed into the rising-flow zone so as not to affect a smelt settling zone communicating with the main and secondary smelt reaction zones and in which slag is separated from matte and metal.
Fig. 2

Sedimentation \( C_1 \times \Sigma \), g/s.m²

Density \( d_1 \), kg/Nm³

- A, \( \dot{V}_1 \)
- M, \( 2 \times \dot{V}_2 \)
- C, \( \dot{V}_2 \)
- M, \( \dot{V}_2 \)
- B, \( \dot{V}_2 \)
- A, \( \dot{V}_3 \)
Fig. 3
Fig. 4

Constructions: A and C
- Sections:
  A: II + III
  C: II

Construction B
- Sections
  B: II + III + IV

Density $d_2$, kg/Nm$^3$

$C_3 \times S_e$, g/s.m$^2$
Fig. 10

Symbols:
- @ Flying dust %
- O Sedimentation g/s-m²

Axes:
- x-axis: Sedimentation la-5e, g-s-m²
- y-axis: Flying dust Sedimentation la-5e, %

Density d₁, g/s-m²
- 1.6
- 1.5
- 1.4
- 1.3
- 1.2
- 1.1
- 1.0
- 0.9
- 0.8
- 0.7
- 0.6

Sheets 10

Patented Aug 19 1975

3,900,310
STABILITY AREAS OF THE
(Fe, Ni, Cu) – S – O – SiO₂ – system

Fig. 11
PROCESS FOR SUSPENSION SMELTING OF FINELY-DIVIDED OXIDE AND/OR SULFIDE ORES AND CONCENTRATES

BACKGROUND OF THE INVENTION

1. Field of the Invention

The invention relates to the field of suspension-smelting of finely-divided oxide and sulfide ores and concentrates.

2. Prior Art

In this connection it must be noted that literature usually gives very little information concerning the solid and smelted materials carried along in a suspension by gas flows in smelting furnaces and other corresponding pyrometallurgical processing units. Furthermore, the available information usually relates to the amount and separation of dusts emerging from the apparatus but very seldom to the behaviour of the sediments of material inside the apparatus and their effect on the process.

In the conventional reverberatory furnaces which are used in mass production of sulfide processing and which can be considered a distant origin of the actual suspension smelting systems, a considerable part is played by dust formation. In order to eliminate dust formation in these apparatuses, hot feeding is replaced by wet feeding which prevents the entrance into the apparatus of very fine, dust-producing material formed particularly in the roasting process. The latest development in reverberatory furnace systems is the so-called Worcra smelting system (West-German Pat. No. 1,533,961 or Canadian Pat. No. 814,026). In order to reduce the amount of dust, this process uses pellets or agglomerates of a concentrate mixture instead of loose concentrates. In regard to the conventional reverberatory furnace smelting, the feeding method has also been changed for the same reason so that the previous feeding along the walls over almost the entire length (60-80%) of the furnace has been replaced by feeding within a zone in the middle part of the furnace usually consisting of less than 30% of the furnace length. In spite of these changes, the amount of dust (flying dust) is still about 3% of the weight of the feed. It can be supposed that, especially in a linear reverberatory furnace type, even a great amount of dust suspended in the gas phase will not, in the form of a furnace sediment, cause considerable increase in the valuable metal content of the slag because the gas phase will carry the dust along when it moves over the feeding bed. The valuable metal content and its increase in reverberatory furnace slag is thus mainly due to the flow phenomena caused by the system itself or by the feeding of additional materials — for example, converter-slag feed. The actual difficulties due to the dissolution of the suspension, dust sedimentation inside the furnace systems, and high amounts of flying dust are created when using actual suspension smelting systems.

In the suspension smelting process developed by the International Nickel Company (U.S. Pat. No. 2,668,107), in which horizontal spraying is used, great amounts of dust are eliminated by adjusting the concentrate granule size suitable for the process so that the granules are considerably large (U.S. Pat. No. 2,668,107, granule size between 95%/65 mesh and 50%/200 mesh, and B.D.P. No. 840,441, granule size between 95%/65 mesh and 5%/200 mesh). Owing to horizontal burning, the amount of dust which sediments inside the system of the said process is so great that even when rough concentrate is used the slag obtained from the furnace is so rich in valuable metals that its after-treatment has been combined with the main process. Matte is produced in the process by suspension smelting of concentrate (a/57.65% Cu and 3.03% Ni, and b/6.95% Cu and 33.77% Ni) during the first operation period — for example, 6 hours. After the removal of the matte, the slag phase (a/0.190% Cu and 0.15% Ni, and b/0.15% Cu and 0.60% Ni) remaining in the furnace is “washed” from the valuable metals by suspension smelting pyrrhotite (a/1% Ni and b/1.25% Ni) during the second operation period — for example, 1-2 hours — and not until then is disposable slag (a/0.35% Cu and 0.16% Ni, and b/0.12% Cu and 0.30% Ni) obtained and, in addition, poor matte as a by-product.

It must particularly be noted that, in spite of the use of rough concentrate, the amount of flying dust in the exhaust gases is very great, about 5% of the amount of feed, even though the gas volumes and obviously also their flowing velocities are very low owing to the pure oxygen or oxygen-rich air used in the burning of concentrate — SO₂ content of the exhaust gases is 85%.

The flash smelting system developed at Outokumpu Oy and its modifications (U.S. Pat. Nos. 2,506,557 and 3,306,708 and Finnish patent application Ser. No. 1000/70 Mar. 29, 1971, U.S. patent application Ser. No. 128,803) include vertical burning of suspended concentrate. In this case, when rough concentrate is used, the solid and smelted materials will not cause furnace sediment problems corresponding to those created in the horizontal process.

The valuable metal contents of the slags of the vertical process are also respectively quite low, and no special aftertreatment is usually needed unless unsuitable additional materials are either fed together with the concentrate or enter the system by other routes. The basic process (Finnish Pat. No. 22,694, U.S. Pat. No. 2,506,557) contains no mention of dusts. In the pyrite treatment process (Finnish Pat. No. 32,465, U.S. Pat. No. 3,306,708), in which sediment dust has no practical importance, the effect of the flying dust amount, about 2-3%, is also technically insignificant. In the known processes for the oxidation and reduction of suspension, the amount of dust in the industrial-scale process is very small in comparison with the used gas amount, or 4-6% of the weight of the feed mixture. It can, however, be noted that the sediments inside the system corresponding to even these small dust amounts, obtained with relatively rough concentrates, have an effect on the valuable metal contents in the slag. However, actual dust problems appear with concentrates more finely-grained than usual; the problems begin only when the vertical suspension can be made to unload only partially. Of new modifications of the suspension process, let us mention the Brittingham process (U.S. Pat. No. 3,460,817) in which an attempt has been made to combine the developed reverberatory furnace process producing raw copper as the final product according to the previously mentioned Worcra process, and the vertical-burning smelting process according to the Outokumpu process. Furthermore, to the system so obtained has been added (U.S. Pat. No. 2,668,107) a sulfide suspension slag wash analogous to the Inco process, but the periodical, shaftless horizontal suspension burning of the latter has been replaced.
by a continuous-working vertical-burning additional shaft. A common flue has been placed between the reaction shaft and the reduction shaft. Thus, the meeting of gas flows from opposite directions ought to make the solid or smelted materials contained in them to fall. There is no mention of the actual product sedimentation rates and flying dust amounts. Because the process has not been applied on an industrial scale, it is difficult to evaluate without any operational information the behavior of the system in regard to these factors.

According to one known process (Canadian Pat. No. 760,925), copper concentrate and additional fuel are injected with the help of compressed air (normal or rich in oxygen) inside smelted ore at a high temperature, at which time the partially oxidized concentrate is arrested in the smelt and smelts forming matte and slag. On the other hand, the additional fuel ought to burn in the smelted material, supplementing the amount of heat required for smelting the ore and, thus, the smelt should always remain at a constant temperature.

According to the process, the copper contents of the mattes are 40.5%, 66.9% and 39.4%, and the copper and sulfur contents of the respective slags 0.33%, 0.52% and 0.35% Cu and 0.20%, 0.23% and 0.26% S. Theoretically, the known balances between the matte and the slag do not materialize with the contents given, especially in regard to sulfur. The obtained result deviates especially from results obtained from reverberatory and flash smelting furnaces. When comparing the copper contents of slags obtained by vertical suspension smelting to the values given above, it can be noted that when the burning of iron is only about 28%, the concentrate and matte contents being 27.8% and about 40% Cu, the values given are by no means rare. It does not give the ferric iron and magnetic contents of the slags so that a comparison with the flash smelting process is without foundation.

On the basis of the balances, no flying dusts are created in this known process, which is very rare in suspension processes. When drafting heat balances on the basis of the material balances, it is noted that practically at the lowest operational temperature that can be considered, 1250°C, the obtained difference between the incoming and outgoing temperatures of the system is zero, so that according to this the used apparatus would have no heat losses (balance references CuO and FeO: into the system: feed mixture 960 kg/824 Mcal and kerosene 40 kg/404 Mcal, or a total of 1227 Mcal; out of the system: matte 635.4 kg/672 Mcal, slag 231.4 kg/80 Mcal, and gas phase (9.4% SO2) 1025 Nm3/474 Mcal, or a total of 1226 Mcal. Difference: 1 Mcal). The heat load values cannot be estimated or compared with other processes because the dimensioning of the apparatus, the delay period, and the capacity values have not been given.

Finally we will review an older suspension roasting process (U.S. Pat. No. 2,209,331), in which dust problems have been discussed more than in the newer processes.

According to the process, materials containing sulfides are roasted or roasted and smelted, and the sulfur content of these materials is recovered preferably in the elemental form or in the form of gases with high SO2 contents. In the process, the roasting is accomplished by dispersing a finely-grained sulfidic material into an oxidizing gas flow which consists of oxygen or oxygen-rich air. After the roasting the material is recovered either in a solid state or in the form of a molten bath. The amount of free oxygen in the oxidizing gas can be regulated so that the free oxygen is used up by oxidizing only part of the sulfur content of the sulfidic material into SO2. The temperature is raised so high that the sulfide smelts in suspension. The iron sulfides have been meant to react to a considerable degree with SO2 to form iron oxide and free sulfur as soon as the oxygen has been used in the formation of SO2.

The processes take place in a vertical reaction tower, in which case the following processes, among others, can be used:

Cocurrent process: A sulfidic material and an oxidizing gas are fed downwards and the product is separated from the gases with the help of a small gas volume and low flowing velocities. The sulfidic material and the oxidizing gas are fed upwards and the velocity is adjusted so that as great a part as possible of the product is carried upward by the gas flow and separated after the tower by known methods. In this case, rough granules will fall countercurrently to the bottom of the tower and are reoxidized when needed.

Countercurrent process: An oxidizing gas flow is fed upwards and the material to be roasted, downwards. The process is particularly suitable for sulfidic material in which the amount of extremely fine particles is not too great.

In certain cases it is recommendable according to the process to divide the sulfidic material into different granule categories and treat each of them separately, and the most finely-grained particles preferably by a parallel flow.

The unloading of the suspension in the process is thus mainly based on the high density of the suspension and very low gas velocities, in which case the settling circumstances mainly determined by the principle of Stokes are obviously achieved especially when using relatively rough particle size distributions.

Even though the described process has mainly been meant for the roasting of iron sulfides, it can, according to the specification, also be used for the treatment of sulfides of valuable metals in the manner described above. Valuable metals Cu, Ni, and Pb are separated from the suspension either as a metal smelt (Cu, Pb), a sulfide smelt, or a powder; in the latter case other known separation processes (crushing, grinding, foaming, magnet separation, chlorination, etc.) are used for the refinement and metal separation.

The object of the present invention is to create a process and apparatus for the suspension smelting of very finely-divided oxide and/or sulfide concentrates by the flash smelting process into sulfide mattes with high or low valuable metal contents, the respective slags being very poor in valuable metals. The process thus also allows for economical processing of finely-divided concentrates by the suspension process, and the main characteristics of the invention are that the remaining suspension flow passing the main reaction zone is fed into the secondary reaction zone for its at least partial dissolution in the smelt before essentially all undissolved remaining suspension is fed into the rising-flow zone in order to prevent the remaining suspension from affecting the smelt settling zone which is communicating at least through the smelt with the main and secondary
smelt reaction zones and in which slag is separated from the matte and metal.

SUMMARY OF THE INVENTION

According to the present invention the residual suspension flow passing the main smelt reaction zone is fed into a secondary reaction zone for its at least partial dissolution in the smelt before essentially all undissolved suspension is fed into the rising-flow zone in order to prevent the remaining suspension from affecting a smelt settling zone which is communicating at least through the smelt with the main and secondary smelt reaction zones and in which slag is separated from the matte and metal.

The process according to the invention can be applied to all pure or mixed concentrates containing Cu, Ni, Co, Zn, Pb, Sn, etc. The smelting system allows for the use of flotation sulphide concentrates with a wide particle size distribution and a very fine average particle size — for example, concentrates obtained from complex and enclosed ores and separated by grinding — which has not been possible by modifications of the suspension process so far. It is now also possible to use corresponding finely-divided, large-surface oxide ores and concentrates containing the said valuable metals, especially if the recently developed techniques of selective suspension sulfidizing are used simultaneously in the process.

BRIEF DESCRIPTION OF THE DRAWINGS

FIG. 1 shows a cross section as seen from the side of a conventional suspension smelting furnace, FIGS. 1A and 1B illustrate modifications made to the basic furnace of FIG. 1, FIGS. 2–4 show graphically the results of sedimentation measurements, FIG. 2 showing the total sedimentation (La-Σ) as a function of the suspension density (d₁) in the lower furnace, FIG. 3 showing the sediment falling in the zone under the reaction shaft (La-RK) as a function of the suspension density (d₂), and FIGS. 4 showing the sediment falling in the other zones of the lower furnace (La-Se) as a function of the suspension density (d₃). FIG. 5 shows the general outline of the smelting installation used in the experiments, FIG. 6 shows a top view of a suspension smelting furnace according to the invention, FIG. 7 shows a section along line C—C in FIG. 9, FIG. 8 shows the copper and sulfur contents of the slag as functions of its magnetic content, FIG. 9 shows graphically the dependence of the reaction velocity constant on the magnetite amount in the shaft product which is in the process of reduction, FIG. 10 shows graphically the sedimentation and flying dust rates as functions of the suspension density (d₁), and FIG. 11 shows the stability ranges of the (Fe, Ni, Cu) –S–O–SiO₂ systems as functions of the sulfur and oxygen materials in the gas phase.

DESCRIPTION OF THE PREFERRED EMBODIMENTS

It can be noted as a general statement that with the growing scale of the flash smelting systems the oxidation and/or reduction of concentrate in a suspension is usually no longer as easy as before when the scale was small, the concentrates were rough, and their granule size distribution was narrow. With a wider size distribution and a smaller particle size, burning, for example, which is a function of particle size, naturally takes place rather unevenly, which causes considerable disadvantages in the general course of the process. On the other hand, suspension formation techniques have developed, and burning velocities have grown after oxygen processes have been put into use. Thus, the level of actual shaft process technology has risen, which naturally makes possible the development of technology in other parts of the system as well.

The effect of concentrate particle size factor in the flash smelting function is discussed below.

In the conventional vertical burning process, the shaft product suspension produced from concentrate unloads under the reaction shaft when the gas flow changes its direction. At this time the bulk of the shaft product — the solid and liquid phases is separated from the gas phase and settled in the main reaction zone under the shaft. Most of the magnetite formed as a product of burning reactions in this main reaction zone with the iron sulfide of the shaft product forming wustite (FeO) which becomes bound to silica, thereby forming a slag from which the valuable metal sulfide smelt, which is poorly soluble in it, is separated as its own phase — matte. The reduction reactions can be only approximately described — iron oxides being taken here as stoichiometric — with the following equations:

\[
(3 + 2x) \text{FeO}_x (s) + \text{FeS}_{3+2x} (s) \rightarrow (10 + 6x) \text{FeO}_x(x+y) + (1-x)\text{SO}_x (g)
\]

\[
(10 + 6x) \text{FeO}_x(x+y) + (5 + 3x)\text{SiO}_2 (s) \rightarrow (5 + 3x) \text{Fe}_2\text{SiO}_4 (s)
\]

The reaction velocity of the magnetite of the main reaction zone can be described by the following equation — on the basis of an empirical process simulation:

\[
d\frac{[\text{SO}_4]}{dt} = K [\text{O}_x]^y [\text{S}_y]
\]

in which Oₓ and Sᵧ are the temporal kinetic activities — approximate strengths — of the oxygen and sulfur involved in the reduction, and K is the reaction velocity constant. At the falling stage of the product of burning the main reaction zone is characterized by a considerable amount of solid [FeOₓ(s) + SiO₂(s)], dependent on the iron content of the matte to be produced, mixed with molten sulfide (Cu, Ni, Co, Fe - S₂), in which case the surface part of the reaction layer in the smelt is naturally characterized by a very high viscosity due to the solid particles. When the magnetite disappears as a result of the reaction reductions and the formed wustite is bound to silicate acid, the amount of solid material decreases, the amount of smelt increases, and its viscosity decreases, at which time the matte and the slag phases rapidly begin to separate from each other owing to the smelt solubility gap prevailing in the system.

In addition to the delay period and the activity, the reaction velocity between the solid and smelted phases is dependent on the contact area of the phases and thereby also on their particle size. In the zone below the reaction shaft, a growing particle size slows down the reduction reactions (the contact surface diminishes, the diffusion distances of oxygen and sulphur grow), and on the other hand the sedimentation rate of the matte through the forming silicate melt grows as a function of the square of the particle size (the sedimenta-
tion rate determined by the principle of Stokes in the limited sense), at which the reduction rate also decreases when the contact possibilities decrease.

However, in the vertical process the reduction reactions usually have time to take place even when relatively rough concentrates are used. On the other hand, when finely-divided concentrates are used, a strong merging of smelt drops occurs in the zone under the shaft so that, in spite of the difference between the products of burning of rough and finely-divided concentrates, the final products in this zone do not greatly differ from each other. When the rest suspension proceeds in the lower furnace from the reaction shaft to the rising shaft, part of its solid and smelted material content settles on the slag surface in the lower furnace. Simultaneously, a strong particle classification takes place in the suspension as a function of the particle size and particle density of the product, in which case the ranges of the lightweight solids and the sulfide smelt separated during the burning will be different. At this time there is a great difference between the behaviour of finely-divided and rough concentrates, which also leads to different changes in the slag phase. The lower furnace sediment formed from rough concentrates is relatively rough, and therefore it usually settles very close to the reaction shaft. The overoxidation of the said shaft product is also very mild so that the sulfide phase of even a continuous sedimentation usually rapidly settles into the matte layer. Owing to the short range, the contact possibilities between the oxide and sulfide particles are good so that the magnetite reduction occurs under advantageous conditions. Because the amount of sediment corresponding to rough concentrate is small, the valuable metal and secondary magnetite contents of the obtained slag phase are low and the growth of these contents is insignificant when proceeding from iron-rich to iron-poor matte.

A suspension containing finely-divided products of burning will not dissolve as easily when its direction is changed. The amount of product which settles in the main reaction zone under the shaft is much smaller than with a rough product of burning under equal conditions and, respectively, the amount of lower furnace sediment is great and quite evenly distributed on the slag surface in the lower furnace. A finely-divided product of burning usually has a wide particle size distribution and, consequently, the particle size distribution of the lower furnace sediment is similar and, in addition, strongly classified as a function of the different densities and particle sizes of the sulfides and solids. Owing to its fine particle size, the lower furnace sediment contains a great deal of overoxidized product and, respectively, less sulfide phase than usual. Though the surface area grows significantly when the particle size decreases, the actual reduction rate of magnetite decreases because, owing partially to the classification (sulfide and oxide phases in different places, etc.) and partially to overoxidation, the kinetic activities of oxygen and sulfur become disadvantageous either quantitatively or otherwise. On the other hand, owing to the small particle size, the sulfides do not sediment at a rate high enough in the slag layer. The said behavior of the sediment results in a rapid increase in the valuable metal content of the slag and an increase in the secondary magnetite amount, which are both independent of each other. When producing high-grade matte from finely-grained concentrates by the conventional process, the result is often the appearance of reduction reaction

\[ \text{Cu}_2\text{S}_{1-x}\text{Fe}_x + 2(1-x)\text{Fe}_2\text{O}_3(s) \approx 2 \text{ Cu} (1) + 6(1-x)\text{FeO} (s, l) + (1-x)\text{SO}_2 (g) \]
due to the overoxidation of the sediment and also its primary low sulfur content. In this case the reduced copper separates as extremely finely-grained and, owing to surface forces, round balls — Cu cloud — with the result that it is impossible for the separated phase to sediment in the matte phase.

Mechanism of the behavior of the suspension to be processed:

The suspension to be processed by flash smelting and containing the feed concentrate and additives is subjected in the system to very many different physical and chemical influences and changes, such as a great change in volume due to the heating of the suspension in the reaction shaft and the corresponding change of velocity; the phase and state inversions of suspended particles due to oxidation reactions and respective changes in the physical and chemical properties of the gas phase; changes and disturbances in the flow due to rapid changes in the direction of the suspension and the geometry of the structure after the reaction shaft; very great changes in the concentration and density of the suspension due to the sedimentation of suspended material, etc. For the above reasons, it is very difficult to estimate the behavior of the suspension. The determination of the surfaces of the local static pressure of flowing suspension as a function of the flowing velocity and the suspension density according to Bernoulli's equation of the conveying systems, and the use of these velocity surfaces prevailing before and after the minimum lines for the determination of the sedimentation rates in the system is only very approximately direct and often practically useless owing to the system and the above factors. Because an analytic solution of the conveying function is not obtainable for the system, variables with the greatest influence and their effect on conveying must be experimentally investigated in each case separately.

The behaviour of the suspension under circumstances corresponding to a flash smelting furnace were first investigated by sample tests and with a pilot-scale furnace, and the results thereby obtained were finally verified with an industrial-scale installation.

Some results of the experiments on the behavior of the suspension are examined below. FIGS. 1, 1A and 1B show a side view of the furnace system involved. The starting point is furnace structure A, known as such. The other structures, B and C, deviate from it in that, while the reaction shaft is the same in all the cases, in case B FIG. 1A the lower furnace has been lengthened and the rising shaft has respectively been moved to position B. In structure C FIG. 1B the lower furnace is the same as in case A, but the rising shaft has been moved to position C. In addition to these, experiments were undertaken with a structure in which the end part of the furnace consisting of the rising shaft was turned to a position perpendicular to the rest of the installation, the area of the second part of the lower furnace corresponding to that in case B. The width of the lower furnace was the same in each structure. Furthermore, it was possible to decrease the normal cross section area of the gas space of the lower furnace by 50%.
For examination of the sedimentation of the suspension, the lower furnace has been divided into zones I, II, III, and IV according to FIG. 1A. In a furnace structure in which rising shaft C has been installed in the immediate vicinity of the reaction shaft, zone I can be called the main reaction zone, zone II the secondary reaction zone, and zone III the settling zone. Furnace C thus deviates from furnace A among other things in that zone II—III has been divided into two zones, II and III, and that the rising shaft has been installed in the immediate vicinity of the reaction shaft in order to minimize the area of zone II. The sedimentation experiments were carried out with concentrates with a particle size of 90%/—400 mesh (surface area 12,000 cm²/cm³). Some experiments were carried out with generally used concentrates with a particle size between values 10%/—400 mesh and 100%/—65 mesh (surface area 2500 cm²/cm³).

The following symbols are used in examination of the sedimentation:

Sedimentation rate: g/s.m²
Total lower furnace sedimentation rate: La—Σ, g/s.m²
La—RK, g/s.m²
Sedimentation rate of the other zones of the lower furnace (or case A: zones II + III, case C: zone II, and cases B and D: zones II + III + IC): La—Se,

Flowing velocity of the suspension: \( v_{i} \), m/s, \( i = 1, 2, 3 \)
Flowing velocity of the suspension in the nozzle of the concentrator dispenser: \( v_{a} \), m/s
Suspension density in the reaction shaft after sulfide burning is \( d_{i} \) and after the sedimentation, La—RK, \( d_{fa} \); density: \( d_{i} \), kg/Nm³

Results of the suspension rate measurements under the circumstances described above I shown in FIGS. 2—4. According to FIG. 2, the total sedimentation rate, La—Σ, is a function of the lower furnace suspension density, \( d_{i} \), in the cases studied. The function is of the form \( La—Σ = q_{i}(d_{i} — d_{i}) \), in which coefficient \( q_{i} \) is a primary function of the lower furnace area and the flowing velocity of the suspension. FIG. 3 illustrates the sedimentation rate, La—RK, in zone I below the reaction shaft. According to the figure it is also a linear function of the suspension density, of the form \( La—RK = q_{i}(d_{i} — d_{i}) \), in which coefficient \( q_{i} \) is, deviating from the previous one, primarily a function of the flowing velocity of the suspension and therefore independent of the geometry of the structure following the reaction shaft under the conditions used.

Owing to the above results, the dependence of the density of the suspension, \( d_{fa} \) after the sedimentation, La—RK, on the post-burning suspension density, \( d_{fa} \), is linear. The function is of the form \( d_{fa} = q_{2}(d_{fa} — d_{fa}) \), in which coefficient \( q_{2} \) is a function of the flowing velocity of the suspension, \( v_{a} \).

Regarding the above observation it must be noted that the obtained sedimentation and density functions are functions of different degrees of the particle size distribution of the suspension, especially with different velocity values. Among other things this can be seen in that the density functions do not run through value zero-zero — value \( d_{fa} \) in the differences — which proves among other things that with different velocities and determined by the velocities, the suspensions contain different amounts of finely-divided particle classes of different size which cannot sediment at all under the conditions used.

The influence of the different flowing velocity values of the suspension (here \( v_{1} > v_{2} > v_{a} \)) on both the sedimentation rate and the \( d_{fa}/d_{fa} \) density distribution values is clear without saying. Reducing into half the cross section area corresponding to the lower furnace gas space (that is, doubling the suspension velocity, \( v_{a} \), to \( 2 \times v_{a} \) in the lower furnace) does not considerably affect the La— rates but it does considerably lower (thus, contrary to the increase in the total flowing velocity) the La—RK rates, as shown by FIG. 3 and the increasing change in the \( d_{fa}/d_{fa} \) distribution.

According to the FIG. 3, the shaft product settling in zones following zone I, La—Se, is no longer a linear function of suspension density \( d_{fa} \) — of density \( d_{fa} \). When the total sedimentation rate of lower furnace zones I and II is examined as a function of the density, \( d_{fa} \), the result is still approximately analogous to that indicated in FIG. 3, but when the number of zones is increased, the result corresponds to data of FIG. 4. After the first zone, the lowering of the density of each suspension as a function of the increase in the area (here also the distance) is initially almost linear, but later deviating when the change in density slows down. The lowerings of the suspension density when using different furnace structures (A, B, C) have different gradients — the steepest with structure C and the least steep with structure B — the gradient difference decreasing with decreasing initial density of the suspension. Thus, it can be noted from the obtained results that, when the lower furnace area following the first zone decreases, a certain surface area value is approached after which decreasing the area no longer increases the sedimentation rate per area. In this case the sedimentation rate is a function of only the suspension density, its granule size distribution, and its velocity. With the suspension here used, this limit area is corresponding to furnace structure A, which can be seen in FIG. 4. When structures B and D, which have equal settler surface areas, were compared with each other, it was noted that the total sedimentation rates, La—Σ, La—RK, and La—Se, were in the latter only slightly higher than in the former. But the sedimentation rates per zone in zones II and III were about 50% lower and in zone IV about 150% higher with structure D than with structure B. The amounts of flying dust were approximately the same with both furnace structures.

A rise in the limit values of the suspension particle size distribution has a very strong increasing effect on sedimentation rates La—Σ and La—RK — FIG. 3. The lower furnace sedimentation rate also becomes much lower so that the amount of product falling in zone II is great (± 80% of product amount La—Se), the other zones containing only little or no sediment. A strong lowering of the flying dust rates occurred when the particle size increased. In the cases observed, the determination of the particle size of the sediment samples as a function of the density of the suspension was carried out by measuring the surface area of the samples. The change in surface area as a function of the respective suspension density was of the same form in the cases observed. When the suspension density decreases, only a slight increase occurs in the surface area of the sediment samples of the first two zones. But a strong gas classification occurs in the remaining zones, while the surface areas per zone increase simultaneously. In fur-
nace structure C, the turning point of the surface area function with a high initial suspension density value is situated at a higher density value than that with a low initial value, that is, the efficiency of gas classification increases when the suspension density decreases. In furnace structure B, the above turning point is situated at suspension density values considerably lower than the previous ones. In structure A, the turning point is between the two previous ones.

Basic principles of the suspension process of finely-grained concentrate are stated below.

On the basis of the above examinations and measurements, the following observations can be made concerning the main prerequisites for the processing of finely-divided concentrates:

As much material as possible must be unloaded from the suspension for the main reaction in the zone under the reaction shaft. This can be achieved, for example, by using a high suspension velocity and a high suspension density in the reaction shaft. The said prerequisites are dependent on the valuable metal content and other properties of the concentrate in each case. It can, however, be generally noted that among other things an increase in the oxygen content of the combustion air, great amounts of feed — that is, activity in the autogenic area, at which time the use of additional fuel is limited — a suitable circulating load in the system (for example, cooling the shaft with return flying dust, in which case the autogenic area contains greater amounts of feed), a high shaft temperature, and the correct dimensioning of the shaft provide means for meeting the said requirements.

The furnace sedimentation which causes slag problems must be minimized by decreasing the sedimentation area and thereby the delay and by allowing, when necessary, a moderate increase — in comparison with the use of rough concentrate — in the amount of flying dust (the total flying dust amount can be diminished by pelleting or agglomerating the return dust). The cross section area of the lower furnace gas space must be optimized to correspond to the highest allowed lower furnace sedimentation rate — a function of, among other things, classification of the product of burning.

The diameter of the rising shaft can be regulated so that the suspension velocities are above the critical conveying velocity, in which case the countercurrent sedimentation of material is small and the danger of growth formation is low.

The absolute value ratio between the flowing velocity of the suspension immediately after the concentrate disperser nozzle and the absolute value of the flowing velocity of the suspension in the reaction shaft varies between the limits 10–100 in different cases.

When adjusting the suspension space and the structural geometry of the furnaces, special care must be taken that the kinetic conditions for the reduction of the magnetite in the shaft product are met while the area of the main reaction zone decreases. The total reaction velocity can also be accelerated by the suspension reduction and sulfidization processes developed by us, although there are other methods, which are usually less effective.

Owing to the character of the process, it can be carried out in several different ways. Processing plants have, however, been designed for the production of relatively rough concentrates and conventional low-grade sulfide mattes. Therefore, for economic reasons, it is possible to investigate in practice only few methods easily applicable to the system. In the investigation of the process, observations, partial studies, product sedimentation rate measurements indicated in the examples, and intermediary smeltings were undertaken with several installations — and naturally, within a very long period of time.

The general outline of the smelting system used in the investigations is shown in FIG. 5. The system has been assembled from conventional, previously known apparatuses. They are the following:

1. Concentrate storage
2. Belt conveyor
3. Dryer
4. Electric filter of the dryer
5. Pneumatic conveyor
6. Redler conveyor
7. Feed tank
8. Concentrate burner
9. Flash furnace
10. Waste heat boiler
11. Heat exchanger
12. Air blower
13. Electric filter
14. Gas blowers
15. Converter
16. Gora furnace
17. Casting of anodes
18. Casting of slag
19. Turbogenerator

When using oxygen-rich air, technical oxygen was added into the air before pre-heating apparatuses II. In the pilot-scale experiments, no heat exchanger was used as a continuation of the radiation part of the boiler to heat the primary air for the smelting, but it was replaced by the normal convection part of the boiler. Neither was the obtained sulfide matte treated further according to the outline in the experiments described. The structural changes in the apparatus for the application of the process according to the invention were carried out in furnace part 9.

A 55 kW intermediate-frequency furnace was used in the intermediary product and other trial smeltings. Heat transmission in the examples in question was obtained by indirect — graphite muffel — heating. The feeding and control apparatuses of the furnaces were conventional. The suspension was usually produced with one concentrate burner. The pre-heating degree of the air used for burning the concentrate was 430 ± 20°C.

The matte, slag, and gas phase samples were studied by conventional methods. The shaft product samples corresponding to the examples were taken just above the smelt surface — in the direction of the center line of the shaft — with an efficiently cooled multisec section sample taking device. The feeding, smelting, and matte-slag separation were carried out as precisely as possible in the manner they take place in an actual furnace unit — that is, the re-smelting effects were minimized. The analyses of the samples and products were made by conventional methods. The determination of ferric iron in the shaft product and the slag phase was made from the oxide phase from which the sulfides had
been separated by the bromine-methanol separation process. 

Products of processing obtained from a conventional furnace structure by shortening the sedimentation area.

A vertical furnace structure provided with a shortened lower furnace secondary reaction zone according to the invention is illustrated in FIGS. 6 and 7. In FIGS. 6 and 7, number 1 refers to the concentrate burner, 2 to the reaction shaft, 3 to the lower furnace, 4 to the rising shaft, 5 to the residual heat vessel, 6 to the heat exchanger, 7 to the cold air flowing into the heat exchanger, 8 to the hot air flowing into the concentrate burner, 9 to ball sooting, 10 to the gas pipe leading to the electric precipitator, and 11 to the fuel for zone reduction. The height of reaction shaft 2 of the vertical smelting furnace was 9.4 m, its diameter 3.8 m, and the cross section area of the shaft 11.64 m². The height of rising shaft 4 was 9.0 m and its diameter 2.75 m. The total area of lower furnace was 85.6 m².

EXAMPLES

In Examples I–VI, the object was to use during the trials the same concentrate mixture (Examples I–III and V relate to the conventional furnace whereas Examples IV and VI relate to a furnace with a shortened lower furnace). It was prepared by mixing concentrates from different mines at a predetermined ratio — if slight deviations occurred, the result was reduced in the examples for the sake of comparison. The average analysis — % by weight — was the following: 20.68 Cu; 33.60 Fe; 0.24 Ni; 0.29 Co; 2.43 Zn; 0.29 Pb; 33.96 S; 4.67 SiO₂; 0.34 CaO; 1.05 MgO and 0.20 Al₂O₃.

The analysis of the concentrate added to the feed mixture was the following: 0.53 Fe; 89.21 SiO₂; 0.27 CaO; 0.35 MgO; and 3.60 Al₂O₃.

Finely-divided concentrate was made from rough concentrate by additional grinding. The composition of the return dust in the feed mixture varied greatly. Especially its oxygen analysis was very inexact numerically. Efforts were made to prevent sulfatizing of flying dusts by precise control of the amounts of leaking air and, when necessary, by a mild reducing burning.

The analyses of the heavy oil and light benzine used for fuel were the following — % by weight:

- Heavy oil: 85.0 C; 11.8 H; 2.5 S; and 0.1 H₂O
- Light benzine: 83.5 C; and 16.5 H

The general objective was to use, other conditions allowing, as high feeding rates as possible in order that the effects of the dust sedimentation and the flying dust be easy to measure. Before the measurements, the furnace installation was always operated until it was in a stationary state, which took about 50–60 hours, depending on the conditions. Balances were drafted of the thermal balance of the installation, especially in regard to reaction shaft 2 (water cooling). The surface temperature of the slag layer in lower furnace 3 was kept at 1300° ± 30°C. The temperature of the matte was respectively 1200°–1260°C, depending on the shaft temperature.

In Examples VII–X refinement of finely-divided — 85%/–/400 mesh — oxidic copper and nickel ores was carried out by the suspension or flash smelting process (Examples VII and IX relate to the conventional furnace whereas Examples VIII and X relate to a furnace with a shortened lower furnace). The used nickel ore was a typical laterite ore containing nickel and chrome, its analysis being the following — % by weight:

- 2.27 Ni; 0.13 Cu; 26.80 Fe; 0.16 Co; 0.80 S; 21.40 SiO₂; 0.03 CaO; 12.27 MgO; 1.70 Al₂O₃. The copper ore was high-grade malachite (about 30% Cu₂O₃-Cu(OH)₂), its analysis being the following — % by weight:

- 18.01 Cu; 18.70 Fe; 1.01 Zn; 30.36 SiO₂; 0.65 CaO; 5.25 MgO; and 4.12 Al₂O₃.

A suspension was produced from each ground ore in the conventional manner, and the suspension was heated to the reaction temperature with oil and air. The oxygen content of the used combustion air was 35% by volume. The sulfidization of the gas phase of the shaft product and the suspension, necessary for high-grade matte and the reduction reactions, was carried out with the help of elemental sulfur and light benzine using zone reduction. The sulfidization reactions were very rapid owing to the small particle size of the ground ore. Because of the high density values of the suspension the zone limits (three zones) were sharp and, thus, the zone volumes small, and therefore the consumption of additional sulfur for the sulfidization of the gas phase was very low.

The reduction of the magnetite of the shaft product in the main reaction zone of lower furnace 3 was controlled by regulating the feeding rates so that the apparent reduction velocities corresponding to normal production were obtained, in other words, there was enough time for the magnetite reduction to take place to a certain degree. The feed mixture and product rates of the experimental operations and the intermediary smeltings and their analyses, as well as the results of the most important observations, are given in Tables 1 and 2 and in FIGS. 2 and 3.

EXAMPLE I

This example represents a normal result of production obtained with rough concentrate under advantageous conditions.

According to Table 2, the results of the shaft sample smelting and of the product were almost the same. The lower furnace sedimentation rate was low, or about 6.5% of the shaft product (or 4.64 g/s.m² — the observation area being 58 m²). Judging from the smelting rates of the shaft product, 95% of the copper of the sediment had time to settle into the matte phase of the lower furnace through the slag layer because of its great drop and particle size — thus, the low copper content of the final slag could be expected. Calculated from the gas delay period, the apparent reaction velocity values of the magnetite reduction are 0.71 — 0.73 × 10⁻³, e⁻s⁻¹ (e = concentration; s = second), which is in accordance with previous experiences when examined per surface unit of the reaction shaft.

EXAMPLE II

In this example, an experimental operation corresponding to that in the previous example was carried out using very finely-divided concentrate.

According to the results, the sedimentation and flying dust rates in the lower furnace have grown considerably, or to 13.2 and 16.0% of the feed mixture. The proportion of sediment in the shaft product is thus about 14.8% or 18.0 g/s.m².

When comparing the intermediary smeltlings with the final result, a strong influence of sedimentation on the final slag (1.8% Cu) can be noted. On the basis of the results, only about 80% of the copper of the sediment
has reached the matte. At the same time, an analysis of the sediment sample proves that its degree of oxidation is higher than that of the actual shaft product. Consequently, the magnetite content of the slag phase has increased, although obviously part of the magnetite phase of the sediment has been reduced under the influence of sulfide matte. The velocity constant obtained for the lower furnace magnetite reduction is $K = 0.71 \times 10^{-7} \text{c}^{-2}\text{s}^{-1}$, which is only about 4% lower than the respective normal value in a reaction system.

EXAMPLE III

This example deviates from those above in that the flying dusts emerging from the system have not been fed back into the system together with the feed mixture. According to the result table, the results do not deviate from those of the previous experimental operation except in regard to the sedimentation and flying dust rates, which are smaller than in the previous ones. The sedimentation rate in the lower furnace was 9.9% of the feed mixture, or about 11.4% of the shaft product, which corresponds to a lower furnace dust load of 7.49 g/s.m$^2$. The lowered dust rates are a consequence of the fact that finely-divided return dust is not contained in the feed — partly it is also a consequence of the lowered Zn and Pb amounts in the feed mixture. Because the particle size of the lower furnace sediment was larger and its amount smaller than previously, the obtained low copper content in the slag was to be expected. Over 80% of the copper content of the sediment settled into the matte phase.

EXAMPLE IV

This example deviates from the previous ones in that the gases and flying dusts are removed from the system by decreasing the area of the secondary reaction zone following the zone under reaction shaft 2 from 58 m$^2$ to about 20 m$^2$, in which case the area obtained for the settling zone following rising shaft 4 is 38 m$^2$.

When comparing this example with Example II, it can be noted from the resulting table that the lower furnace sedimentation rate has decreased considerably, from 13.2% to 4.2%. The obtained sedimentation rate corresponds to about 4.7% of the shaft product, or a dust load of 14.61 g/s.m$^2$. The flying dust rate has risen from 16.0% to 18.4% of the feed mixture, which can be considered a relatively small rise.

Owing to the low sedimentation rate, the copper content of the slag has lowered considerably — 0.51% — in comparison with Example II — 1.78% Cu. The ferric iron content in the shaft product falling directly under reaction shaft 2 is almost the same in each case, but the magnetite content in the slag has decreased from 15.9%, which is the value in Example II, to 10.4%.

Judging from the obtained results, over 90% of the copper contained in the sediment has settled into the matte phase. This amount is higher than was supposed for the reason that only the roughest and at the same time fastest sedimenting part of the residual suspension has had time to fall in the lower furnace. The value obtained for the magnetite reduction velocity constant corresponding to the gas delay is $K = 0.75 \times 10^{-8} \text{c}^{-2}\text{s}^{-1}$, which is corresponding to those ordinarily obtained.

EXAMPLE V

In the experimental operation according to this example, high-grade copper matte was produced by using finely-grained concentrate and a conventional furnace system. In order to eliminate reaction shaft wall effect in this and the following case, deviating from the previous ones, a less homogeneous suspension was used in concentrate burning and a content of 1% free oxygen was allowed in the combustion gases. It can be noted from the results in Table 2 that, in comparison with the production of low-grade matte, the sulfur content of the shaft product has decreased considerably. In this case, the overoxidation of the most finely-divided part of the concentrate mixture forming sediment and dusts is natural. Because there was a considerable amount of lower furnace sediment with the used feed mixture and operation method (or about 14% of the shaft product), which corresponds to a dust load of 12.4 g/s.m$^2$, the increase in the copper and ferric iron contents in the final slag phase could be expected. When examining the obtained results, it can be noted that only about 60% of the copper of the sediment has ended up in the final matte. About 65% of the copper content in the slag phase is in a metallic form, which also indicates a strong participation of copper sulfide in the magnetite reduction after the amount of sulfidic iron has decreased sufficiently. Let it be mentioned that even less advantageous slag values than these have been obtained in similar experimental operations, in which cases the copper content has been 4–5%, magnetite content 18–30%, and sulfur content less than 0.1% — let it be noted that in spite of the great variation in the oxidation circumstances, no oxides or ferrites of copper in solid state have been observed in the slags.

The obtained gas delay period value for the apparent reaction velocity constant of the lower furnace magnetite reduction is $K = 1.17 \times 10^{-8} \text{c}^{-2}\text{s}^{-1}$, which is slightly higher than normal.

EXAMPLE VI

This example deviates from the previous one in that the reaction gases and dusts were removed from the system in a manner corresponding to that in Example IV. In addition, in this case the reduction of the shaft suspension in zones was carried out by using three petroleum ether nozzles 11 (fitted below the middle of reaction shaft 2 symmetrically and at the same height) in order to increase the low amount of sulfate after the burning of the shaft product and to increase its activity. In comparison with the previous examples, the shaft product in the present case is almost of the same grade, and only slightly richer matte than previously is obtained when it is smelted. It is practically impossible to obtain a representative shaft sample after the zone reduction. Thus, the average analysis obtained from the sediment samples of lower furnace 3 is quite inexact — Table 2. A probable shaft product — 2 — corresponding to the final products has been calculated from the final products and the said analysis. In comparison with Example V — 14% ~12.4 g/s.m$^2$ — the furnace sedimentation rate is low —4%, which corresponds to a surface load of 10.6 g/s.m$^2$. Respectively, according to the balance values the flying dust rate has grown only 20%; the low value is also partially caused by the inhomogeneous suspension. The low magnetite and copper contents in slag according to Table 2 are, however, possible with the used technology only when using a zone reduction of the suspension.

Before the light benzine reduction, the rate of gas emerging from the reaction shaft was about 21 080
N\text{m}^3/\text{h} and its dry analysis — % by volume — was the following: 14.3\ SO_2; 1.0\ O_2; 0.2\ CO_2; 84.5\ N_2. Petroleum ether was fed into each of the three shaft zones at the rate of 50 kg/h. The following gas analysis values — % by volume — were obtained from the mean of gas samples centrally crossing the zone of the lower part of reaction shaft 2 (the values being inexact because there was still a rather great \( P_0 \) gradient in the direction of the diameter): 1.7\ H_2; 3.8\ CO; 7.4\ CO_2; 4.6\ SO_2; 82.5\ N_2. The oxygen pressure obtained from ratio CO_2/CO was \( P_0 = 10^{-8} \) atm. Reduction zone size determination at the same point gave a zone area which was approximately 10 % of the cross section area of the shaft. The total gas rate calculated from the above values is 21\ 410\ N\text{m}^3/\text{h}. Calculating from the gas delay periods, apparent values \( K = 1.17 - 1.18 \times 10^{-3}\ \text{cm}^3/\text{cm}^2\text{s}^1 \) are obtained for the magnetite reduction velocity constant, the values corresponding to normal values.

An examination of the heat balance of reaction shaft 2 shows that benzine reduction did not cause a notable change in the mean temperatures of the reaction products.

**EXAMPLES VII–VIII**

In the experimental operations according to these examples, the heating, reduction, and sulfidizing of oxides in suspension were carried out so that the products obtained were a high-grade sulfide matte and a slag poor in valuable metals. The examples deviate from each other in regard to the sedimentation zone areas of lower furnace 3 so that Example VII represents a conventional furnace system and Example VIII a system (FIG. 6–9) in which the settling area after the main reduction zone is about 20\ m^2. The suspension process was the same in both cases. Oxide ore was heated while slightly reducing it so that the suspension temperature before zone-sulfidization was 1450\°C and the oxygen pressure of the respective gas phase was \( P_0 = 10^{-7} \) atm. The volume of the suspension was then 6390\ N\text{m}^3/\text{h}. The volume of the three sulfidization zones was 12.5% of the suspension volume. After sulfidization and additional reduction, the zone gas-volume was 960\ N\text{m}^3/\text{h}, the final temperature and oxygen pressure of the suspension \( T = 1400\°C \) and \( P_0 = 1.0 \times 10^{-6} \) atm. After shaft product sulfidization the composition of the gas phase — % by dry volume — was the following: 4.4\ H_2; 0.7\ H_2S; 11.7\ CO; 22.4\ CO_2; 0.1\ CO; 3.7\ SO_2; 57.1\ N_2. According to Table 1, the decrease in the area of the secondary smelt reaction zone had an effect analogous to the results of the previous experiments, in other words, the proportional amount of sediment decreased greatly, but its rate per area remained almost unchanged. From material balances — Table 2 — it can be noted that the lowering of the sedimentation rates had an advantageous effect on the nickel and copper contents in the slag. In slags obtained from processing copper concentrate, the said effect is greater than the one obtained here. According to observations, this is mainly due to the difference between nickel- and copper-containing sulfides in regard to, for example, surface qualities. It must be noted in general that the separation of nickel-containing sulfides from the slag phase and their settling through the slag phase is faster than that of copper-containing sulfides. Because usually the nickel present in slags is bound to the oxide and silicate phases, the nickel content in the slag is mainly determined by its degree of oxidation. In the case discussed, the nickel content in slag is small and corresponds well to the \( Fe^{2+}/Fe^{3+} \) ratio in the slag — the apparently small ferric iron content corresponds to the low total iron content in the slag.

**EXAMPLES IX and X**

The experimental operations according to these examples were carried out with malachite ore, but in other respects they are completely analogous to Example VII and VIII.

The temperature of the heating and pre-reduction phase of the shaft process was 1450\°C, and the oxygen pressure of the gas phase (11\ 940\ N\text{m}^3/\text{h}) was \( P_0 = 10^{-4} \) atm. Owing to the high copper content in the ore, the need of zone reduction was greater than in the previous examples, or about 36% of the gas phase. After sulfidization and reduction, the gas volume of the zone was 5480\ N\text{m}^3, its temperature 1400\°C, its oxygen pressure corresponding to value \( P_0 = 1.1 \times 10^{-5} \) atm, and the analysis of the gas phase the following — % by volume: 4.8\ H_2; 0.5\ H_2S; 13.1\ CO; 26.4\ CO_2; 0.1\ CO; 2.9\ SO_2; 52.2\ N_2. According to Tables 1 and 2, the lower furnace sedimentation rates and their effects on the final products correspond to those of the previous experiments. Especially notable among the results in the increasing effect of finely-divided sediment on the copper content in the slag when a conventional lower furnace structure is used in a case in which the amount of ferric iron would correspond to a very low copper content in the slag (Example IX). The apparent reaction velocity values of magnetite reduction in the example fall between values \( k = 0.73 - 0.75 \times 10^{-5}\ \text{cm}^2/\text{s} \). Though the obtained values are low when high-grade matte is produced, they do correspond to observations made during production. Because of the low iron content in the feed mixture the amount of magnetite to be reduced is low and the reduction mechanism thus mainly corresponds to that in the production of low-grade matte.

The most important results of the examples can be seen in FIGS. 8–11. FIG. 8 shows the copper and sulfur contents in slag as a function of its magnetite content. According to the figure, the copper content in slag is a regularly continuing function of its magnetite content. The obtained result is, however, only apparent and characteristic of the used conditions and, thus, not a balance function. In practice the function is influenced by the grade of matte (because of the process, the valuable metal content gradient in the slag-matte system is vertical so that the slag is in balance with a lower-grade matte than that produced on industrial scale), the magnetite reduction velocity, sedimentation rate, suspension burning technology, etc. Often, however, a regular \( Cu - Fe^{2+}/Fe^{3+} \) function is obtained which indicates the dynamic balance of the flash smelting system (but there is no dependence between, for example, the valence of the copper in slag — 0, 1, 2 — and the said dependence under the used circumstances). According to the figure, the sulfur content in slag is not a very regular function of its magnetic content. When producing high-grade matte, however, a dependence between a low sulfur, a high ferric iron, and a high copper content in slag can be observed regularly—FeS_2 is used up and Cu,S begins to participate in the reduction; this dependence can be eliminated by a partial reduction of the shaft suspension. The dependence of the reaction velocity constant as a function of the amount of magne-
tite in shaft product under the process of reduction in
dicated in partial FIG. 9 has been used in the example
and in trials in general as one means for observing the
reality of the obtained results (in other words, the re-
sults of this highly complicated process mechanism are
not a consequence of random factors). Thus, observa-
tions prove that the reaction mechanism and kinetic
prerequisites of the magnetic reduction of this modifi-
cation of vertical suspension smelting are controlled,
for example, in the experimental operations according
to the examples. The used process is not theoretical.
The sedimentation and flying dust rates shown in
FIG. 10 indicate very clearly the extent and direction
of the behavioral changes in the suspension and their
harmony with the previous observations, although the
arrangements for the trials were very simple for eco-
nomical production reasons.
FIG. 11 shows the stability ranges of the metal, sul-
fide, oxide, and silicate phases which can be considered
when using zone-reducing and sulfidization as a func-
tion of the sulfur and oxygen pressures within the used
temperature range; they were calculated in connection
with the observations.

According to the stability graph, the composition of
the gas phase after zone reduction in the production of
rich copper matte according to Example VI is within
the stability range of molten copper and iron sulfides.
The composition of the basic gas phase (10g P_o =
2.00 and 10g P_s = 12.75) is outside the coordinates.
The pre-heating and pre-reduction gas phase of nickel-
containing laterite — Example VII — is within the sta-
bility range of the magnetite and nickel, nickel oxide,
and nickel olivine phases, but the gas phase corre-
spanding to zone reduction — Example VIII — is
within the range of nickel and iron sulfides, as it should
be. The pre-reduction process for the refinement of
malachite — Examples IX and X — was carried out
until its oxygen pressure was lower than that obtained
in laterite reduction but otherwise the obtained result
is analogous to the result of the nickel process.

It can be noted in general that it is easy to produce
a suspension of solid material in a gas volume which is
at a low temperature and therefore rather small. On
the other hand, the dissolution of suspension at a high
temperature, at which time its viscosity values are often
manifold and the density of the suspended material
low, is a very difficult task, especially when the sus-
pered material is very finely-divided. According to
observations, the amount of material separated from
the suspension in the main smelt reaction zone in-
creases as a function of the suspension density and the
suspension velocity (the increase being faster than
would be expected on the basis of the change in den-
sity. The sedimentation rate of the other zones in
the lower furnace also increases as a function of the velo-
city and the suspension density. This sedimentation,
under influence of which the valuable metal and mag-
netite contents in the slag phase increase, is very harm-
ful to the process, especially when producing high-
grade mattes. When the grade of concentrate improves,
the overoxidation of fine particle classes, effective clas-
sification, the decrease in the sedimentation rates of
matteslag separation, and the decrease in the reaction
velocity of reduction increase the harmful effect of the
furnace sedimentation leading to uneconomical after-
treatment processes of slag when using the conven-
tional vertical process to produce low-grade mattes.
In the present description, a process has been intro-
duced in which the disadvantageous effects of sedimen-
tation can be decreased decisively by regulating the
sedimentation rate (La—Se) in the zones following the
reaction shaft. In this case, when using a conventional
rising shaft, a greater than usual amount of returnable
flying dust must be allowed in the process, or when
using a system without a rising shaft, the suspension ve-
locity after the concentrate disperser must be in-
creased. This increase in the suspension velocity short-
en the time used for actual sulfide oxidation when a
conventionally dimensioned reaction shaft is used. For
this reason, depending on the type of concentrate and
other factors, the height of the reaction shaft must be
increased and simultaneously the area of the main
smelt reaction zone (which is a function of the cross
section area of the reaction shaft) must be kept large
enough to obtain a sufficient delay period for the re-
duction of the magnetite in the shaft product. On the
other hand, this process also makes it possible to use
finely-divided concentrates and to carry out the pro-
cess by using the suspension process. Thus, it makes
possible large production units while eliminating the
previous difficulties and uneconomical after-treatment
of slag.
On the basis of investigations, the flying dust load
mentioned above can be reduced considerably by ag-
gglomerating or micropelletting of flying dust to be fed
back into the process. The composition of the flying
dust corresponds approximately to the product of burn-
ing in the reaction shaft — any differences can further
be evens out by a mild rising shaft reduction so that
a shaft process is not necessary. The treated return fly-
ing dust can be fed into the smelt reaction zone either
together with the feed mixture, in which case its tem-
perature can be increased simultaneously, or by using
some other feeding method depending on the system.
By this method, the effect increasing the sedimentation
of very finely-divided material and flying dust is elimi-
nated and, when desired, and improved slagging of, for
example, zinc and lead is obtained — in Example III, an
effect analogous to the method described above is ob-
tained by not feeding flying dusts back into the system.
The effect of the suspension density and velocity on
the sedimentation rates is noteworthy. With present-
day technology, by increasing the oxygen content in the
burning air, sufficiently high and advantageous suspen-
sion density values can be easily obtained, but at the
same time the suspension velocity values are propor-
tionately lowered within the same system. Decreasing
the reaction area in order to increase the suspension
velocity is mainly dependent on the valuable metal con-
tent of the initial suspension. The reduction velocity of
the magnetite of the main smelt reaction zone deter-
mines the dimensions of the system very precisely so
that often the said process is not practicable at all.

According to experience, the reduction velocity of
magnetite increases with improved oxide-sulfide con-
tact so that a decrease in the plasticity of the colli-
sion surface has an advantageous effect on the process.
Structurally the plasticity of the collision surface can be
influenced so that a slanted bottom structure deepening
towards the matte-slag separation zone is placed
immediately under the smelt surface in the part of the
lower furnace containing the main and secondary zones
(for example, FIG. 1B, structure C, zones I and II). Thus, in an arrangement corresponding to furnace
structure C, the viscous flow of material from under the reaction shaft is mixed with the sediment, in which case the sediment contains a great amount of sulfide phase in relation to the oxide phase (when needed, the amount of sulfide can be increased by an intermediary reduction, thereby ensuring the viscous flow and eliminating the effect of the sedimentation). In addition to the increase in the reduction velocity and the drop size, the above solutions are advantageous especially when mass-producing high-grade mattes in the selective sulfidization of overoxidized valuable metal components (Ni, Co, etc.) of the shaft product in the lower furnace reaction zone (for example, when pentlandite process:

\[(\text{Fe}, \text{Ni})_x\text{S}_y \rightarrow \text{NiO}, (\text{Fe}, \text{Ni})_x\text{S}_y, \text{Fe}_2\text{O}_3, \text{FeO}, \text{SO}_2; \text{NiO} + y\text{FeS} \rightarrow x\text{O}_2 y\text{S}_z (\text{Ni}_x\text{S}_y), \text{FeO}, \text{SO}_2).\]

Only linear furnaces have been discussed in the description of the process so far. This is mainly due to the fact that changing the shape of the furnace will usually not make the process notably more advantageous because of sediment classification. The use of non-linear furnace structures in production often causes great practical difficulties and wearing phenomena due to growths and "pile formation" of unreacted sediment. The objective in the present examination has also been to carry out the experimental operations and measurements under conditions in which the dissolution of the suspension on the wall surfaces of the reaction shaft has been eliminated. With the "turbulent spray" type concentrate burners developed by us it is possible, when producing high-grade matte and metal in particular — especially when using oxygen-rich combustion air — to burn the concentrate and to conduct the products of burning to the wall of the reaction shaft, in which case the magnetite in the product of burning will be reduced effectively under the influence of the good oxide-sulfide contact, the increased reaction surface, and the lengthened delay period. At the same time the process decreases the furnace sedimentation and flying dust rates. At the present technological level, however, the disadvantages of the method are so obvious (great wear and tear, increased heat losses, etc.) that using it in the conventional process has been avoided (in addition, it must be noted that when using finely-divided concentrates, the obtained decrease in the sedimentation and flying dust rates is not yet sufficient). On a small scale, an experimental model of a semilinear furnace structure has also been tested in which the part of the lower furnace connecting the reaction and rising shafts — that is, the secondary reaction zone — forms part of the circular ring or segment from which the linear matte-slag separation part begins. In this case, an unusually great lower furnace sediment amount obtained when using (by using central forces) to the autogenically lined lower furnace wall, at which time the drop merging and reduction velocities increase and simultaneously the sediment is conducted away without causing slag damage. The method has a disadvantage in that the furnace size is still limited in order to obtain a sufficient sediment separation. On the other hand, the method provides a special advantage — especially when it is combined with reaction shaft wall reactions — when reaction

\[\text{Cu}_x\text{S}_y + (1 + x)\text{Fe}_2\text{O}_3(s) \rightarrow \text{Cu}(1 + x)\text{FeO} (s_1) + (1 + x)\text{SO}_2(g)\]

must be induced in connection with the production of copper, especially at a relatively low temperature, because the drop size of the formed metal can then be sufficiently increased and, respectively, a "copper cloud" which is for the most part finely-ground is eliminated from the slags. Note that it is possible to apply either a resistance heating or electric arc apparatus — or both — for satisfying the need of heat in the lower furnace matte-slag separation zone when necessary.

### Table 1

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<th>Example Balance component</th>
<th>Quality</th>
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<th>III</th>
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<th>VI</th>
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### Table 2.

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<th>Fe²⁺</th>
<th>Me</th>
<th>S</th>
<th>O</th>
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<td>Example I</td>
<td>Feed mixture</td>
<td>1000.0</td>
<td>18.24</td>
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<td>4.09</td>
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<td>(10.69)</td>
<td>(17.03)</td>
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<td>Sedimentation</td>
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<td>(3.76)</td>
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<td>303.9</td>
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<td>(3.77)</td>
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<td>69.4</td>
<td>18.0</td>
<td>28.4</td>
<td>(16.2)</td>
<td>20.4</td>
<td>7.0</td>
<td>(14)</td>
<td>8.0</td>
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| Example II | Feed mixture | 1000.0 | 18.31 | 29.12 | 2.54 | 5.15 | 26.30 | 1.9 | 14.26 |
| Shaft product | 100.0 | 21.06 | 34.21 | 16.16 | 3.31 | 11.57 | (9.82) | 16.84 |
| Slug-1 | 41.69 | 50.04 | 22.18 | (4.52) | 2.24 | 22.77 | 2.59 | 12.75 |
| Slug-2 | 56.0 | 50.05 | 44.58 | 4.34 | 4.24 | 1.69 | (13.6) | 30.00 |
| Sedimentation | 131.5 | 21.3 | 34.5 | 19.5 | 3.4 | 8.9 | (12) | 17.0 |
| Slug-3 | 295.8 | 49.35 | 22.65 | (4.63) | 2.30 | 22.79 | 2.65 | 0.09 |
| Slug-4 | 411.4 | 1.78 | 44.25 | 7.65 | 4.21 | 1.43 | (14.3) | 29.73 |
| Flying dust | 160.1 | 18.6 | 26.3 | (15.5) | 17.1 | 7.0 | (12) | 12.5 |

| Example III | Feed mixture | 1000.0 | 18.25 | 29.66 | 0.1 | 2.87 | 29.97 | 0.0 | 14.60 |
| Shaft product | 100.0 | 21.15 | 34.76 | 16.24 | 2.01 | 11.75 | (9.63) | 17.28 |
| Slug-1 | 41.73 | 50.00 | 22.69 | (4.26) | 1.60 | 23.02 | 2.44 | 0.15 |
| Sedimentation | 56.06 | 21.3 | 45.13 | 5.13 | 2.40 | 1.84 | (13.4) | 30.72 |
| Slug-2 | 98.9 | 21.3 | 35.0 | 18.4 | 2.0 | 10.1 | (11) | 17.4 |
| Slug-3 | 420.1 | 0.58 | 45.13 | 6.51 | 2.40 | 1.60 | (13.8) | 30.72 |
| Flying dust | 120.0 | 20.0 | 30.0 | (18.1) | 11.3 | 7.9 | (12) | 13.7 |
| Balance component | Amount | Cu | Weighted analyses | Fe | Fe<sup>2+</sup> | Me | S | O | SiO<sub>2</sub>
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<td>12 (21.0)</td>
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**TABLE 2.—Continued**

| Balance component | Amount | Cu | Weighted analyses | Fe | Fe<sup>2+</sup> | Me | S | O | SiO<sub>2</sub>
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<td>(11)</td>
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**What is claimed is:**

1. An improved process for the suspension smelting of finely-divided oxide or sulfide ores and concentrates within primary and secondary reaction zones, wherein a suspension of finely-divided raw material in preheated air, oxygen or both is fed downwards at a temperature of at least the reaction temperature of a primary reaction zone formed by the suspension and the smelt underneath to oxidize and partially smelt the suspension raw material, whereinafter the oxidized raw ma-
terial is partially sulfidized and the suspension flow is deflected perpendicularly so that the bulk of the raw material in the suspension flow impinges against the surface of the accumulated smelt in a main smelt reaction zone under the primary reaction zone and the remaining suspension flow is fed into a rising-flow zone, and the solids to be returned to the reaction zone are separated from the residual suspension flow, the improvement comprising feeding the remaining suspension flow passing the primary reaction zone into the secondary reaction zone so that its partial dissolution in the smelt in said secondary reaction zone does not exceed five percent by weight of the total shaft product before essentially all undissolved remaining suspension is fed into the rising-flow zone in order to prevent the remaining suspension from affecting a smelt settling zone which is communicating at least through the smelt with the main and secondary smelt reaction zones and in which slag is separated from the matte and metal.

2. The improved process of claim 1, wherein the suspension is produced from a raw material 90% of which is finely-divided with 400 mesh as its largest particle size.

3. The improved process of claim 1, wherein the entrance of the remaining suspension into the smelt settling zone is prevented by keeping the velocity of the residual suspension in the area of the secondary reaction and rising-flow zones above the critical conveying velocity.

4. The improved process of claim 1, wherein the ratio of the absolute values of the flowing velocity of suspension after the concentrate disperser nozzle to the flowing velocity of suspension in the reaction shaft is kept within the range of 10 to 100.